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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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MILLING METHODS AND COSTS AT THE GOLDEN CYCLE MILL, COLORADO SPRINGS, COLO.

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BY

L. S. Harner

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MILLING METHODS AND COSTS AT THE GOLDEN CYCLE MILL, COLORADO SPRINGS, COLO.[1]

By L. S. Harner[2]

INTRODUCTION

This paper which describes the milling practice at the Golden Cycle mill is one of a series of similar papers being prepared by the United States Bureau of Mines.

The mill of the Golden Cycle Corporation, formerly known as the Golden Cycle Mining and Reduction Co., is located at Colorado Springs, Colo., and has been operated since 1907. It is now and has been from its inception almost exclusively a custom mill.

HISTORY

The original mill had capacity to treat 37,000 tons of oxidized silicious ores per month by roasting and cyanide methods. When it treated what may be termed "basic ores" the capacity was 32,000 tons per month.

A small-capacity cyanide unit was added to the original plant about 2 years ago for the purpose of treating miscellaneous gold-silver ores which did not require roasting. In the latter part of 1929 a concentrator unit was added for the treatment of complex sulphide ores by selective flotation methods. The tailings of this latter unit are further treated in the cyanide plant for additional recoveries of gold and silver.

The changes in operating methods at the Golden Cycle plant since 1907 have been made because of--

(a) Additional operating knowledge gained from direct experience and from information received from outside sources.

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[1] The Bureau of Mines will welcome reprinting of this article provided the following footnote acknowledgment is made: "Reprinted from U. S. Bureau of Mines Information Circular 6739."

[2] Manager, Golden Cycle Corporation, and one of the consulting engineers, U. S. Bureau of Mines.

(b) Mechanical improvements which include materials of better quality furnished by manufacturing companies.

(c) Changes in methods and equipment brought about by changes in the character of ores treated.

The development of milling methods at this plant cannot be shown by cost and recovery figures over the entire period of plant operation. The milling practices that yielded high recoveries and low costs when the plant started operations would result in low extractions and very high costs if practiced on the present type of ore treated. As the character of ore changed, new methods and new expensive and more efficient equipment were necessary if operations were to continue. At all times changes were preceded by experimental work.

ORE TREATED

The gold ores which are treated by roasting and cyanide methods are the sulphotelluride ores received almost exclusively from the Cripple Creek district. These ores are chiefly gold bearing and contain very small amounts of silver and negligible quantities of base metals such as lead, copper, zinc, arsenic, antimony, and mercury.

Gold and silver ores treated by direct cyanide methods are received in small shipments from quite widely scattered districts.

Complex sulphide ores which are treated by selective flotation methods are furnished in large part by the well-known old mining districts of Clear Creek and Gilpin Counties, the Alma, Leadville, Creede and San Juan areas.

The Cripple Creek ores received by the mill are classified either as silicious or basic ores; the former are obtained from the upper levels or so-called "oxidized" and "leached" zones of the mines and the latter from below the oxidized zones. The siliceous oxidized ores can be readily crushed, ground, and roasted, whereas the denser, harder, and slicker ores obtained below the oxidized zone give more difficulty in the crushing, grinding, and roasting operations. The tabulation which follows presents typical analyses of these two classes of ore.

Typical analyses of Cripple Creek ores milled

Constituents, percent	Siliceous oxidized ore	Dense hard ore
Insoluble	86.70	75.90
Al_2O_3	2.30	3.40
Fe	3.50	4.00
CaO	1.57	5.12
S	1.79	1.8 to 2.30
MgO	0.50	1.40
Loss ignition	3.20	6.50

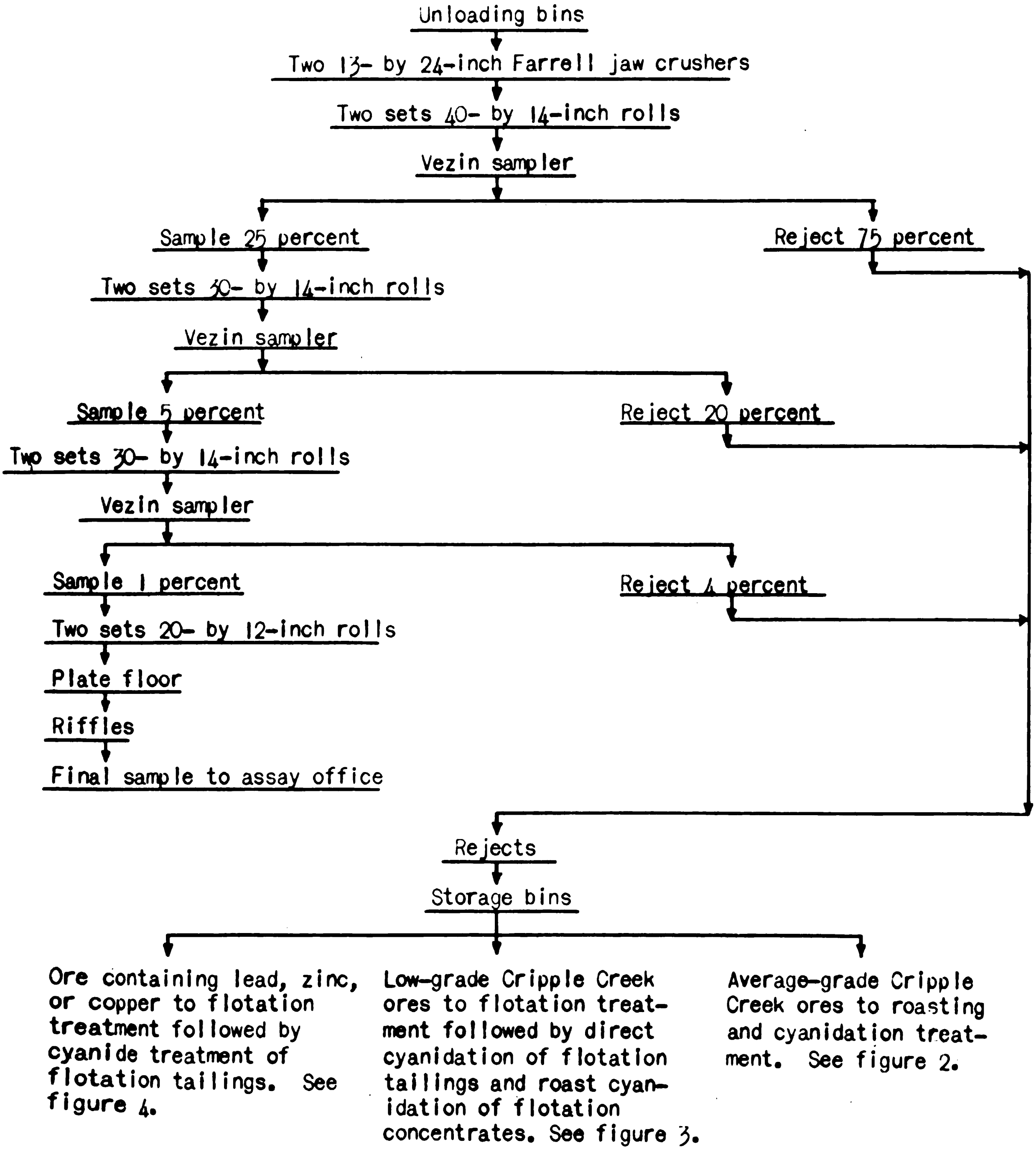
Unloading bins
Two 13- by 24-inch Farrell jaw crushers
Two sets 40- by 14-inch rolls
Vezin sampler
Sample 25 percent
Reject 75 percent
Two sets 30- by 14-inch rolls
Vezin sampler
Sample 5 percent
Reject 20 percent
Two sets 30- by 14-inch rolls
Vezin sampler
Sample 1 percent
Reject 4 percent
Two sets 20- by 12-inch rolls
Plate floor
Riffles
Final sample to assay office
Rejects
Storage bins
Ore containing lead, zinc, or copper to flotation treatment followed by cyanide treatment of flotation tailings. See figure 4.
Low-grade Cripple Creek ores to flotation treatment followed by direct cyanidation of flotation tailings and roast cyanidation of flotation concentrates. See figure 3.
Average-grade Cripple Creek ores to roasting and cyanidation treatment. See figure 2.

Figure 1.- Flow sheet of sampling mill.

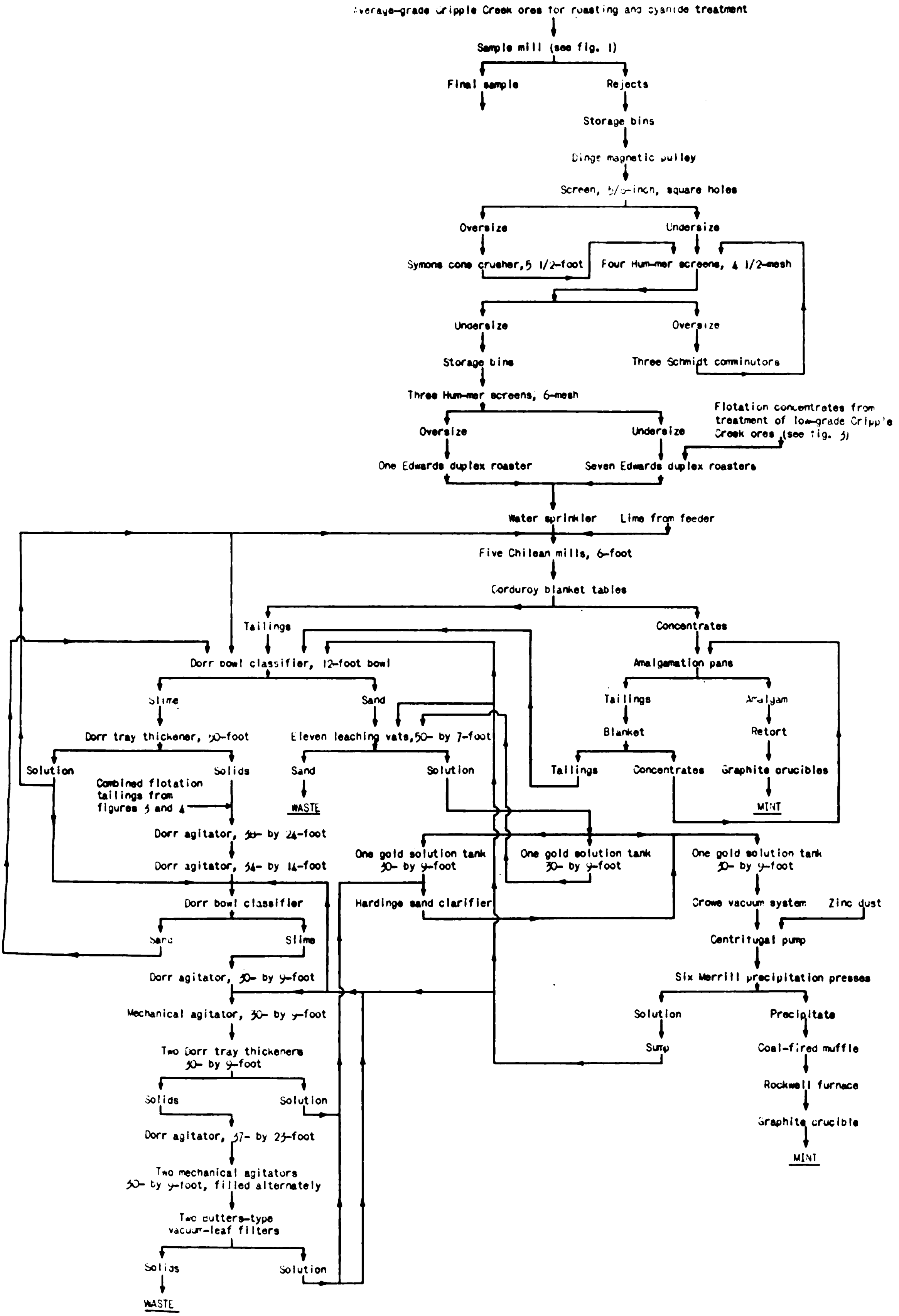
Average-grade Cripple Creek ores for roasting and cyanide treatment
Sample mill (see fig. 1)
Final sample
Rejects
Storage bins
Dings magnetic pulley
Screen, 5/8-inch, square holes
Oversize
Undersize
Symons cone crusher, 5 1/2-foot
Four Hum-mer screens, 4 1/2-mesh
Undersize
Oversize
Storage bins
Three Schmidt comminutors
Three Hum-mer screens, 6-mesh
Flotation concentrates from treatment of low-grade Cripple Creek ores (see fig. 3)
Oversize
Undersize
One Edwards duplex roaster
Seven Edwards duplex roasters
Water sprinkler
Lime from feeder
Five Chilean mills, 6-foot
Corduroy blanket tables
Tailings
Concentrates
Dorr bowl classifier, 12-foot bowl
Amalgamation pans
Slime
Sand
Tailings
Amalgam
Dorr tray thickener, 50-foot
Eleven leaching vats, 50- by 7-foot
Blanket
Retort
Solution
Solids
Sand
Solution
Tailings
Concentrates
Graphite crucibles
Combined flotation tailings from figures 3 and 4
WASTE
MINT
Dorr agitator, 38- by 24-foot
One gold solution tank 30- by 9-foot
One gold solution tank 30- by 9-foot
One gold solution tank 30- by 9-foot
Dorr agitator, 34- by 14-foot
Dorr bowl classifier
Hardinge sand clarifier
Crowe vacuum system
Zinc dust
Sand
Slime
Centrifugal pump
Dorr agitator, 30- by 9-foot
Six Merrill precipitation presses
Mechanical agitator, 30- by 9-foot
Solution
Precipitate
Two Dorr tray thickeners 30- by 9-foot
Sump
Coal-fired muffle
Rockwell furnace
Solids
Solution
Graphite crucible
Dorr agitator, 37- by 23-foot
MINT
Two mechanical agitators 30- by 9-foot, filled alternately
Two Butters-type vacuum-leaf filters
Solids
Solution
WASTE

Figure 2.- Flow sheet of treatment for average-grade Cripple Creek ores.

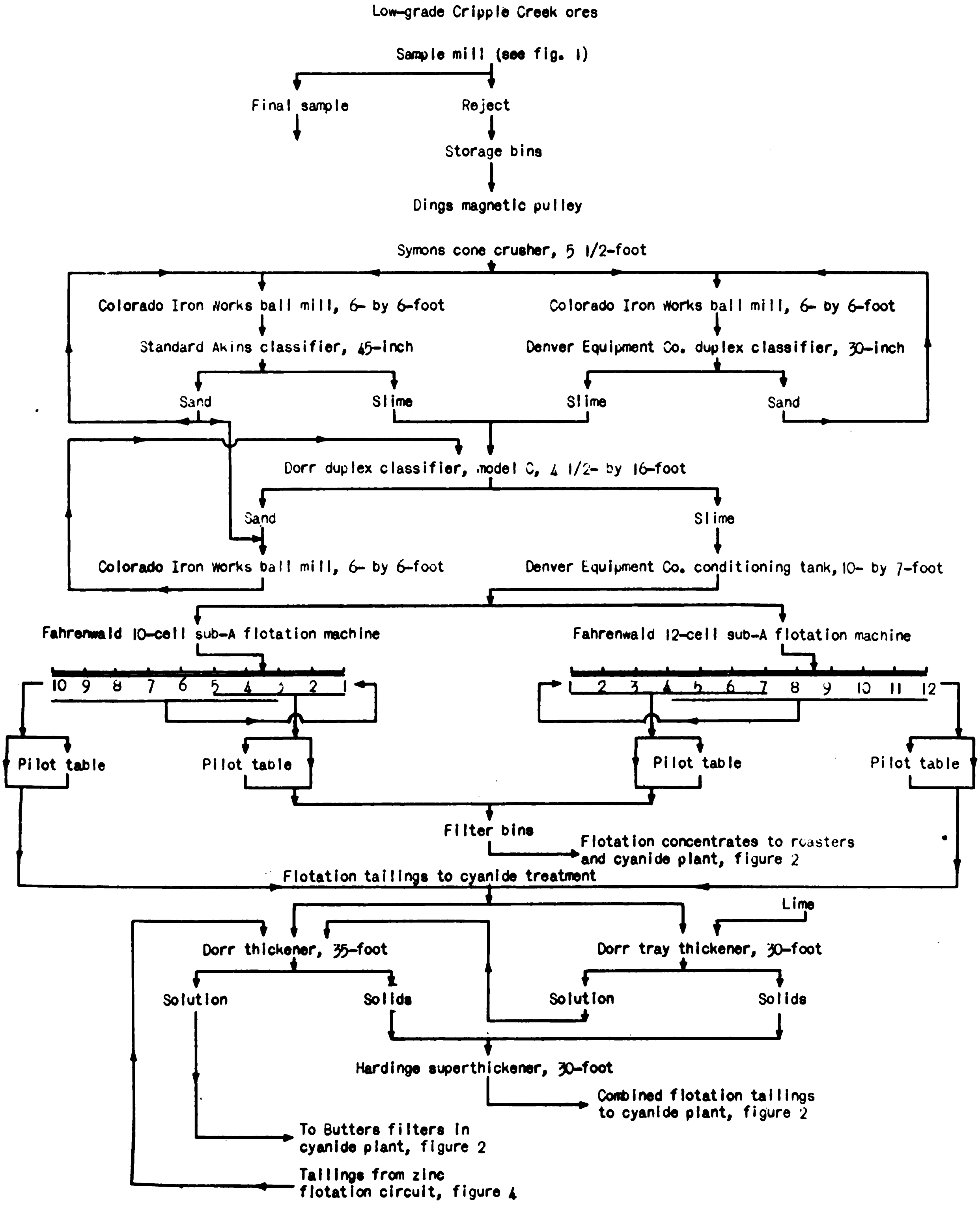
Low-grade Cripple Creek ores
Sample mill (see fig. 1)
Final sample
Reject
Storage bins
Dings magnetic pulley
Symons cone crusher, 5 1/2-foot
Colorado Iron Works ball mill, 6- by 6-foot
Colorado Iron Works ball mill, 6- by 6-foot
Standard Akins classifier, 45-inch
Denver Equipment Co. duplex classifier, 30-inch
Sand
Slime
Slime
Sand
Dorr duplex classifier, model C, 4 1/2- by 16-foot
Sand
Slime
Colorado Iron Works ball mill, 6- by 6-foot
Denver Equipment Co. conditioning tank, 10- by 7-foot
Fahrenwald 10-cell sub-A flotation machine
Fahrenwald 12-cell sub-A flotation machine
10 9 8 7 6 5 4 3 2 1
1 2 3 4 5 6 7 8 9 10 11 12
Pilot table
Pilot table
Pilot table
Pilot table
Filter bins
Flotation concentrates to roasters and cyanide plant, figure 2
Flotation tailings to cyanide treatment
Lime
Dorr thickener, 35-foot
Dorr tray thickener, 30-foot
Solution
Solids
Solution
Solids
Hardinge superthickener, 30-foot
Combined flotation tailings to cyanide plant, figure 2
To Butters filters in cyanide plant, figure 2
Tailings from zinc flotation circuit, figure 4

Figure 3.- Flow sheet of treatment for low-grade Cripple Creek ores.

The slight physical and chemical changes that take place in the ores after mining and before milling have no appreciable effect on ore treatment.

WATER AND POWER

Water is pumped from a near-by creek and is stored in seven large tanks.

Power is chiefly supplied from the company's steam electrical plant which is located at the mill. The city of Colorado Springs furnishes any additional power required.

PRESENT METHODS OF TREATMENT

Sampling

As previously mentioned, the Golden Cycle plant operates chiefly on custom ores and is therefore equipped with a complete sampling unit. Figure 1 shows the flow sheet of the sampling plant.

From the unloading bins the ore is fed to two 13- by 24-inch Farrell jaw crushers. The crushed product is further reduced by two sets of 40- by 14-inch rolls and then passed to a Vezin sample cutter which takes a 25 percent cut. The sample is again reduced by two sets of 30- by 14-inch rolls and then passes to a second Vezin sample cutter which removes a 20 percent cut. The sample is further reduced by 30- by 14-inch rolls and passed to a third Vezin sampler which takes a 20 percent cut. This sample which amounts to 1 percent of the original material is crushed by two sets of 20- by 12-inch rolls and further reduced in size by riffles. The final sample is dried, ground, and sent to the assay office. The rejects from sampling operations, which constitute the plant feed, are conveyed to storage bins.

Classification of Treatment Methods

The plant as previously noted treats three general classes of ores by the methods which follow:

1. The average-grade Cripple Creek gold ores are treated by roasting followed by cyanidation. Figure 2 presents a flow sheet of this treatment.

2. Low-grade Cripple Creek gold ores are first treated by flotation as shown in figure 3. The flotation concentrates are sent to the roasters which treat the average-grade Cripple Creek ores, and the roaster calcines follow the treatment outlined in figure 2. The flotation tailings are cyanided directly as shown in figure 3.

3. Ores which contain lead, copper, or zinc are treated by selective flotation methods as indicated in figure 4. The lead concentrates and zinc concentrates produced are shipped to smelting points; the flotation tailings of the zinc circuit are cyanided directly as shown in figure 3.

This paper does not describe the details of flotation treatment as practiced at the Golden Cycle plant but deals mainly with the roasting and cyanide treatment presented in figure 2.

TREATMENT OF AVERAGE-GRADE CRIPPLE CREEK ORES

Referring to figure 2, the ore is conveyed from the storage bins on a 14-inch belt which is equipped with a Ding's magnetic head pulley for the removal of tramp iron. The conveyor belt discharges the material onto a stationary screen having 5/8-inch holes and set at a slope of about 40°. The oversize passes to a 5 1/2-foot Symons cone crusher set at 3/8 inch; the crushed product joins the stationary screen undersize and both are fed to four Hum-mer screens equipped with 4 1/2-mesh Rek-Tang screen cloth. The Hum-mer screen oversize is further reduced by three Schmidt comminuters; the latter are dry-grinding ball mills. These mills operate in closed circuit with the 4 1/2-mesh screens. The undersize products of the 4 1/2-mesh Hum-mer screens comprise the feed to the roasters and are conveyed to storage bins.

The Symons cone crusher will handle 100 tons of ore per hour, the feed being 3 1/2- to 4-inch maximum size and the product 1/2-inch maximum size.

Ores received in the early operating days of the Golden Cycle plant could be reduced to roasting size quite readily. However, as lower levels in the mine were reached the character of the ore changed to a more basic variety and at times to somewhat dolomitic unoxidized material. The crushing of these lower-level ores became more difficult.

At the beginning of operations the crusher product was reduced to roaster feed size by three 6- by 6-foot Schmidt comminuters. The latter used 5-inch balls and were equipped with diagonally slotted screens having 9/64- by 1/2-inch holes. These screens were attached to each mill and operated in closed circuit with it, the screen oversize being returned to the mill.

As the difficulty of crushing ore increased a 36-inch Symons horizontal disk crusher and a 48-inch Symons vertical disk crusher were added to the crushing circuit ahead of the Schmidt ball mills. The remodeled unit operated in a satisfactory manner for a while, but as more of the mines reached deeper levels the cost of crushing ore increased.

The next change made was the addition of a 10-foot by 48-inch dry-grinding Hardinge ball mill to the grinding units already installed. This change was followed by the installation of a 5 1/2-foot Symons cone crusher.

The Symons cone crusher displaced the horizontal and vertical Symons disk machines and also rendered the operation of the 10-foot Hardinge mill unnecessary. Since the installation of the cone crusher, power costs in this department have been reduced one half and the time required for crushing has been reduced one third. The saving made by this installation amounts to $0.08 per ton of ore handled.

Roasting

The ore is delivered from the roaster storage bins to three Hum-mer screens equipped with 6-mesh Rek-Tang screen cloth. Table 1 gives a screen analysis of the feed to the Hum-mer screens.

The screen oversize is roasted in one Edwards duplex furnace;[3] the screen undersize, with concentrates from the flotation unit handling low-grade Cripple Creek ores, is distributed to seven Edwards duplex roasters. Each roaster will handle from 115 to 130 tons of ore per 24 hours. Screen analyses of coarse and fine roaster feeds are presented in table 1.

The Edwards roasting furnaces are 115 feet long by 13 feet wide and each has an active hearth area of 1,495 square feet. A cooling hearth, 44 feet long and 13 feet wide, adjoins each roaster at the discharge end. Roaster hearths have a slope of 2 inches per foot.

Each roaster is equipped with 54 revolving rabbles which move the ore from the feed to the discharge end of the furnace. The rabble arms of the roasting hearth revolve at a speed of 3 r.p.m. and those of the cooling hearth at 6 r.p.m. Each roaster requires 16 hp. for operation; ore passes through the roaster in the average time of 6 hours.

The rabble arms of the hearths are water cooled. Cooling water used per roaster amounts to 90 to 110 gallons per minute depending on the temperature of the hearth and the rate of feeding ore. The water enters the rabble arms at a temperature of from 23° to 33° C. and after leaving the rabbles, is sent to a cooling pond and from there is pumped to storage tanks for reuse.

Colorado Springs lignite coal of the following analysis is used for roaster fuel:

Water	per cent	20.20
Volatile matter	do.	54.65
Fixed carbon	do.	19.75
Ash	do.	5.10
Sulphur	do.	0.30
Heating value	B.t.u. per pound	8,700

The fire box in use is of the Western Fire Box Co. type; it is a semigas producer and uses live steam. The consumption of coal in the producer at present ranges from 230 to 240 pounds per ton of ore roasted, and an additional 8 percent is required to generate the steam for the producer. Sizing of the ore before roasting has been found to reduce the coal required from 75 to 100 pounds per ton of ore when the roasters operate at a temperature of from 800° to 900°C. Lower roasting temperatures are used at times and under these conditions the saving of coal by sizing the ore is not so pronounced.

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[3] Hofman, H. O., General Metallurgy: McGraw-Hill Book Co., New York, 1st ed., 1913, p. 424.

Extensive tests to determine roaster-stack losses were made in 1909. Gas velocities and gas pressures were determined and dust samples were caught in filter bags. At the beginning of these tests the loss of metal in stack-dust was found to be $0.13 per ton of ore. This loss was reduced to $0.04 per ton of ore by lowering the feed discharge point into the roasters, by providing better draft control and by choking the calcine discharge.

Temperatures are controlled by Brown indicating pyrometers; recording pyrometers are also in use for plotting temperature curves over long periods. Sulphur analyses are made each day.

The roasting of the basic ores may result in the production of two very undesirable compounds; namely, CaS and $CaSO_4$.

When CaS is present in the roaster calcine it consumes a large amount of cyanide in the leaching operation; it is also a reducing agent, and acting in this manner decreases the dissolution rate of the gold and at times decreases this rate to a point where gold ceases to dissolve. If silver is present in solution CaS will precipitate it and may prevent its final recovery.

Calcium sulphate, if present, crystallizes in pipe lines, launders, and tanks; it chokes filter and clarifier mats; it coats zinc-dust cloths and frames or settles out in zinc boxes, if the latter are in use; finally it coats zinc shavings or dust and thereby inhibits precipitation.

Due to the difficulties described, it is advisable to roast the ore so as to produce as small amounts of these two compounds as possible. It is difficult, however, to approach complete elimination of the sulphur in the case of the basic ores. Considerable quantities of $CaSO_4$ are usually present in the calcines but troublesome quantities of CaS may be largely avoided.

A study of the behavior of $CaSO_4$ at elevated temperatures with certain fluxes has been made by Hofman and Mostowitsch and most of the statements which follow on this subject have been taken from their report.[4]

When calcareous ores containing metallic sulphides are roasted the lime is converted almost quantitatively to $CaSO_4$, providing sufficient sulphur is present. When the $CaSO_4$, however, is heated in the presence of either CO or C, which happens in roasting operations, a reduction of the $CaSO_4$ takes place as indicated by the reactions which follow:

Reduction of $CaSO_4$ by CO: $CaSO_4 + 4\ CO = CaS + 4\ CO_2$.

This reaction is slow below 680°C., is rapid between 750° and 850°C. and is practically completed at 900°C. The reaction also take place without loss of sulphur.

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[4] Hofman, H. O., and Mostowitsch, W., The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes: Trans. Am. Inst. Min. Eng., vol. 39, 1909, p. 628.

Reduction of $CaSO_4$ with C: $CaSO_4 + 4\ C = CaS + 4\ CO$.
$CaSO_4 + 2\ C = CaS + 2\ CO_2$.

These reactions are fairly rapid at 700°C. and are completed at 1,000°C.

At the Golden Cycle plant CaS is formed as noted in these reactions. The carbon monoxide is introduced into the roaster with the fuel and the solid carbon from the cars which deliver ore to the plant, as many of these cars are used for the transportation of lignite coal to the mines. In addition to the solid carbon, introduced as noted, additional carbonaceous cinder from locomotives is introduced into the cars enroute to the plant.

This solid carbonaceous material if not completely burned during roasting of the ore forms a substance analogous to charcoal which readily precipitates gold in leaching operations. A roasting temperature high enough to completely burn this material also decomposes calcium and magnesium carbonates. The CO_2 gas liberated surrounds the ore particles for a time and prevents the oxidation of any CaS present to $CaSO_4$. Later during the roast the CaS is partly converted to $CaSO_4$ and CaO but some CaS remains in the final calcines. During leaching operations the small amount of CaS which goes into solution is soon oxidized or precipitated by zinc or iron salts. No satisfactory method has yet been found to prevent the formation of $CaSO_4$ during roasting or to remove it cheaply after it has deposited in various parts of the equipment.

The calcines drop from the cooling hearth of the roaster onto a reciprocating drag conveyor, 350 feet long. The hot ore is moved along the drag by sheet-iron fins 34 inches long by 8 inches deep. The conveyor requires 16 1/2 hp. for operation. The drag discharges the calcines to a conveyor belt which is specially prepared to handle hot material. The product when reaching this belt is at a temperature of about 85°C. and contains 2.5 percent of moisture; the water is added as a spray between the drag and belt conveyors.

Grinding

The belt conveyor delivers the calcine to five 6-foot Chilean mills for grinding. The Chilean mills are equipped with screens which have 0.0496-inch openings, and the grinding is done in cyanide solution.

Recovery of Coarse Gold

The pulp from the Chilean mills is fed to blanket tables 16 by 12 feet in size for the recovery of the so-called "coarse gold." Each table is divided into four sections with a 3-inch drop between sections. The blankets used are either ordinary cheap part wool, fairly lightweight cotton, or corduroy; they are changed twice in 24 hours. The concentrates caught by the tables are removed by stretching the blankets over a rail and washing with cyanide solution which is delivered by a pressure hose. The recovery of gold by these tables amounts to between 22 and 40 percent of the total gold.

Amalgamation

The concentrates washed from the blankets are directed into a small hopper-bottomed receiver and from there are fed to a small grinding pan. Mercury is added to the pan and the coarse free gold recovered by amalgamation. Since the amount of concentrates to be treated is small they are fed to the grinding pan at a slow rate and most of the minerals other than gold are ground fine enough by the mullers to overflow from the pan. The overflow pulp is added to the general mill circuit.

The amalgam is periodically drained through a plug located near the bottom of the pan and after cleaning is squeezed through a cloth which resembles bed ticking. The amalgam is retorted and the mercury returned to the mill for reuse. Fouling of mercury during pan amalgamation of concentrates is negligible; the average yearly loss of mercury for the past 3 years amounted to 275 pounds.

The retort bullion is melted and cast into bars for shipment to the mint at Denver. The gold bullion varies from 920 to 940 fineness in gold and from 45 to 55 fineness in silver.

Leaching of Sand

The tailings of the blanket tables and the overflow pulp of the grinding pan are delivered to a Dorr bowl classifier by centrifugal pumps. The classifier is equipped with a 12-foot-diameter bowl and is operated with liberal amounts of back-wash water added to the raking compartment; this results in the production of a sand practically free from slime. These sands amount to about 70 percent of the mill feed and contain from 22 to 26 percent of moisture. They are conveyed and distributed to the sand leaching vats by a rubber belt conveyor.

There are 11 leaching vats 50 feet in diameter and 7 feet deep; each will hold 600 tons of dry sand. The time of leaching varies from 6 to 8 days. The charges are drained and aerated four to five times during the leaching cycle for 8-hour periods.

The solution which overflows the tanks during charging and the solution drained during the first 48 hours of leaching flows into gold-solution storage tanks. From the storage tanks it is fed continuously to the precipitation presses.

After the 48-hour initial leaching period barren sump solution from the precipitation presses is used for leaching and washing purposes. These solutions upon leaving the leaching vats are either sent to the Chilean mills where the ore, as previously mentioned, is ground in cyanide solution or, if the solution is low grade, it may be pumped to one or two of the last tanks for leaching purposes. The leaching with barren sump solution is followed by a final water wash.

The leached sand charge is removed from vats by sluicing with water into launders which lead to a common sump. From the sump the tailings are pumped in three stages to the tailings dump. A charge of 600 tons is removed from a vat in about 3 1/2 hours by two men operating a 2 1/2-inch hose equipped with a 3/4-inch nozzle. Water at 200 pounds pressure is used.

The tabulation which follows presents a screen-assay analysis of the sand tailings from leaching operations.

Screen size; mesh	Weight, per cent	Gold assay, ounces per ton
Plus 20	18	0.01
Plus 30	28	.01
Plus 40	14	.01
Plus 60	25	.01
Plus 100	6	.02
Plus 150	5	.02
Plus 200	2	.03
Minus 200	2	.03
Composite	100	0.012

Leaching of Slime

The overflow pulp of the Dorr bowl classifier contains about 7 percent of solids. This pulp is thickened to 42 percent of solids in a 50-foot Dorr tray thickener, the overflow solution being returned to the Chilean grinding mills. The thickened pulp is discharged by four Dorrco diaphram pumps and is delivered to a 38- by 24-foot continuous Dorr agitator. The overflow from this agitator feeds another continuous Dorr agitator 34 by 14 feet in size. This agitator discharges into a Dorr bowl classifier, the sand returning to the first classifier and the overflow slime feeding a 30- by 9-foot continuous Dorr agitator which in turn overflows into a 30- by 9-foot continuous agitator of the paddle type. The pulp leaving here is rethickened to 50 percent of solids in two 30- by 9-foot Dorr tray thickeners. The overflow solutions from these thickeners go to a clarifier and from there to storage precipitation tanks.

Thickened pulp from the Dorr tray thickeners is pumped to a 37- by 23-foot continuous Dorr agitator which is followed by either of two 30- by 9-foot mechanical agitators. These latter agitators feed the Butters filters.

The total time of agitation amounts to between 65 and 75 hours. The solids handled contain 4 to 5 percent 100-mesh laboratory screen oversize and about 84 percent minus 200-mesh material.

Filtering

The filter equipment comprises two Butters intermittent type of filters each equipped with 87 canvas-covered 5- by 9 1/2-foot frames. The filters are fed from the agitators as noted, the agitators being filled alternately.

The slime cake is given a barren-solution wash which is followed by a short water wash. The tailings from the filter presses contain 0.0276 ounce of gold per ton of which $0.0245 is soluble gold. The filtered solutions pass to a clarifier and from there to the precipitation presses.

Clarification of Solutions

Clarification at this plant is best effected by means of the Hardinge sand clarifier; this applies to solutions from the slime plant before precipitation. The clarifier used is 30 feet in diameter and has a capacity of 3,000 tons of solution per day. It is equipped with a sand filter bed which is kept clean by a spiral scraper that discharges the sludge in a manner similar to that of an ordinary thickener.

Precipitation

Zinc shavings contained in zinc boxes were formerly used for the precipitation of gold from cyanide solutions. Since 1929, precipitation has been effected with zinc-dust and Merrill presses have been used for the recovery of the precipitate. The solutions are passed through Crowe vacuum equipment before precipitation. The barren solution, which contains 0.02 ounce of gold per ton, flows into sumps and is used as wash solution for both sand and slime treatments.

At times it has been found difficult to satisfactorily precipitate the values from the cyanide solutions derived from the treatment of roasted Cripple Creek ores. The factors which follow have been found to influence the results of precipitation: (a) Type of ore, whether basic or otherwise; (b) quantities of sulphates and sulphides formed during roasting operations; (c) rate of feed to roasters, and (d) temperature used in roasting.

The addition of lead acetate, in amounts depending upon conditions in roasting, has been found at times to be beneficial.

Treatment of Precipitates

When zinc shavings were used for precipitation the zinc boxes were cleaned up by removing the contents from the first or first and second compartments of the seven compartment boxes. The precipitate was treated with sulphuric acid in a tank. When most of the zinc had dissolved, the sludge was dropped into a small air-tight tank and from there forced by air pressure into a small clean-up press. The gold sludge was washed in this press with hot water for the removal of zinc sulphate. The press was then dismantled, the sludge cake removed, and charged into a muffle type furnace. The muffle was brought to a red heat and most of the lead and any remaining zinc oxidized. The roasted product was mixed with flux in the proportion of 100 pounds of residue to 84 pounds of flux. The flux contained 28 pounds of silicious sand, 28 pounds of bicarbonate of soda, 14 pounds of borax glass, and 14 pounds of fluorspar. The charge, contained in graphite crucibles, was melted in oil-fired tilting furnaces.

When the change from zinc shavings to zinc-dust was made the sulphuric acid treatment was discontinued. The precipitate, after removal from the Merrill press, is now put into iron pans and mixed, while wet, with sodium nitrate in the proportion of 125 pounds of precipitate to 25 pounds of nitrate. The pans are placed in a furnace equipped with cast-iron muffles and the charge is heated to a red heat. The heating results in the drying and sintering of the charge which may be subsequently handled without loss of dust. The bullion is then mixed with the sand-borax glass-soda nitrate-fluorspar flux, previously noted, and charged to a fire-clay tile lined Monarch-Rockwell type of furnace for melting.

METALLURGICAL DATA AND RECOVERY

Screen analyses of various plant intermediate and final products for the year 1929 are given in table 1. Metallurgical data for the year 1929 are presented in table 2. A summary of metallurgical results for 1929 which includes distribution of products, assays, and recoveries is given in table 3.

The recovery of gold at this plant has been found to be affected by the conditions which follow:

(1) Very careful attention to the details of the roasting of basic ores is necessary.

(2) The complete removal of free gold particles of any appreciable size by blankets, or some other means, is essential if tailings of low gold content are to be obtained within reasonable time periods.

(3) The consumption of chemicals, the formation of calcium sulphate and other cementing compounds and the amounts of deleterious sulphides produced vary with the types of ores and with the manner of roasting.

(4) Recovery of gold has at times decreased even when the roast appears satisfactory, and this condition is apparently due to the accumulation of zinc salts in the cyanide solutions. Sufficient amounts of sulphides are usually present in the calcines to precipitate injurious soluble zinc; but when the recovery decreases, the addition of soluble sulphides is in general beneficial.

(5) As a general condition, both sand and slimes are difficult to wash. At the beginning of washing operations satisfactory displacement of gold-bearing solution apparently takes place but later in the washing operation the dissolved gold appears to be removed chiefly by diffusion rather than displacement.

It has been found that the use of what might be termed "excessive barren washes" for the removal of dissolved values in both sands and slimes is advantageous.

(6) In a plant of this character mechanical leaks of ore or solution and waste of values of any kind should be very carefully watched.

COSTS

A summary of plant costs for 1929 is presented in table 4. The distributions of labor and power are given in tables 5 and 6, respectively.

Table 1. - Screen analyses of plant products for 1929

Screen size	Weights, percent									
	Feed to 40-inch rolls	Feed to Symons cone crusher	Feed to comminuters	Feed to 6-mesh Hum-mer screens	Roaster feeds		Feed to Chilean mills	Dorr bowl classifier		
					Coarse	Fine		Feed	Sand	Slime
Plus 3-inch, square	17.2	---	---	---	---	---	---	---	---	---
Plus 2-inch, square	15.5	7.7	---	---	---	---	---	---	---	---
Plus 1-inch, square	28.0	42.2	---	---	---	---	---	---	---	---
Plus 13/16 inch, square.	4.7	13.2	---	---	---	---	---	---	---	---
Plus 9/16 inch, square..	5.4	15.7	---	---	---	---	---	---	---	---
Plus 3/8 inch, square...	6.6	13.2	6.2	---	---	---	---	---	---	---
Plus 5/16 inch, square .	3.0	3.1	11.8	---	---	---	---	---	---	---
Plus 8 mesh	7.8	2.8	76.1	14.7	95.7	4.0	14.7	---	---	---
Plus 10 mesh	2.1	.2	1.9	16.0	4.3	17.5	16.0	---	---	---
Plus 14 mesh	.9	.1	.8	10.6	---	12.0	10.6	---	---	---
Plus 16 mesh	.4	.2	.2	2.1	---	2.4	2.1	---	---	---
Plus 20 mesh	1.0	.2	.3	7.6	---	8.6	7.6	12.6	18.0	---
Plus 30 mesh	1.6	.2	.2	11.0	---	12.5	11.0	19.6	28.0	---
Plus 40 mesh	.7	.1	.1	7.5	---	8.5	7.5	9.8	14.0	---
Plus 60 mesh	2.4	.6	1.4	11.0	---	12.4	11.0	17.5	25.0	---
Plus 80 mesh	.1	.1	---	.7	---	.8	.7	---	---	0.1
Plus 100 mesh	.7	.1	.1	2.5	---	2.8	2.5	5.5	6.0	4.1
Plus 150 mesh	.6	.1	.1	3.1	---	3.5	3.1	4.9	5.0	4.8
Plus 200 mesh	.7	.1	.3	2.8	---	3.2	2.8	3.5	2.0	7.0
Minus 200 mesh	.6	.1	.5	10.4	---	11.8	10.4	26.6	2.0	84.0
Totals	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 2. - Metallurgical data for 1929

Total ore treated	tons	298,500
Days operated		365
Operating time per day	hours	24
Operating time	percent	100
Average ore treated per 24 hours	tons	817.8
Assay of plant heads	ounces gold per ton	.502
Recoveries of gold:		
By amalgamation	percent	25.33
By cyanidation before classification	do.	55.10
Cyanide sand leaching	do.	10.70
Cyanide slime leaching	do.	5.68
Combined recovery	do.	96.81
Losses of soluble gold in tailings:		
Sand tailings	per ton	$0.0132
Slime tailings	do.	$ .0245
Average	do.	$ .0166
Consumption of reagents, pounds per ton of ore treated:		
Cyanide, 100 per cent sodium cyanide		.964
Hydrated lime		6.119
Zinc-dust		.269
Hydrochloric acid		.169
Lead acetate		.037
Strength of solutions, pounds sodium cyanide per ton of solution		2.0 to 0.7
Consumption of balls	pounds per ton of ore treated	.378
Consumption of liners	do.	.105
Consumption of Chilean mill tires and dies	do.	.053
Net water consumption, including power plant, gallons per ton of ore treated		1,045

Table 3. - Summary of metallurgical results for 1929

	Moisture, percent	Dry weight, tons	Dry weight, percent of total	Assay, ounces gold per ton	Weight of gold, ounces	Percent of total gold	Percent of total gold	
							Recovered	Lost
Plant heads	3.7	298,500	100.00	0.502	149,847	100.00	---	--
Heads to roasters ..	3.6	298,500	100.00	.502	149,847	100.00	---	--
Roaster calcines ...	---	285,366	95.60	.525	149,817	99.98	---	0.02
Amalgamation	---	---	---	---	37,956	25.33	25.33	--
Cyanidation during amalgamation	---	---	---	---	82,571	55.10	---	--
Dorr bowl classifier heads	---	285,366	95.60	.1026	29,290	19.55	---	--
Heads to sand leaching	26.2	199,756	66.92	.0922	18,418	12.29	---	--
Solutions from sand leaching	---	---	---	---	16,041	10.70	---	--
Tailings from sand leaching	11.2	199,752	66.92	.0119	2,377	1.59	---	--
Heads to slime leaching	93.1	85,610	28.68	.1270	10,872	7.26	---	--
Solutions from slime leaching	---	---	---	---	8,509	5.68	---	--
Tailings from slime leaching	32.8	85,608	28.68	.0276	2,363	1.58	---	--
Combined plant tailings	17.7	285,360	95.60	.0166	4,737	3.17	---	3.17
Combined cyanide solutions	---	---	---	---	107,121	71.48	71.48	--
Totals	---	---	---	---	---	---	96.81	3.19

Table 4. - Summary of plant costs for 1929

	Operating labor	Supervision	Power[1]	Supplies	Reagents	Repairs		Miscellaneous	Total
						Labor	Other		
Unloading, operations and repairs combined...	$0.0680	---	---	$0.0030	---	---	---	---	$0.0710
Sampling, including jaw crushers and rolls...	.0576	---	$0.0099	.0181	---	$0.0114	$0.0146	$0.0274	.1390
Symons crusher	.0095	---	.0112	.0364	---	.0059	.0004	---	.0634
Comminuters	.0143	---	.0379	.0268	---	.0161	.0206	---	.1157
Screens and elevators	.0079	---	.0049	.0047	---	.0030	.0001	---	.0206
Roasters	.1333	---	.0464	.3501	---	.0172	.0325	.0556	.6351
Chilean mills	.0161	---	.0644	.0259	---	.0141	.0044	---	.1249
Conveying	.0520	---	.0321	.0194	---	.0191	.0087	---	.1313
Amalgamation, operations and repairs combined	.0150	---	.0013	.0060	---	---	---	---	.0223
Classification	.0035	---	.0010	.0011	---	.0017	.0010	---	.0083
Sand leaching	.0287	---	.0035	.0072	---	.0042	.0039	---	.0475
Slime leaching	.0142	---	.0156	.0061	---	.0035	.0035	.0156	.0585
Filters	.0152	---	.0052	.0120	---	.0039	.0028	---	.0391
Clarifier	.0061	---	.0013	.0033	---	.0002	.0001	---	.0110
Precipitation	.0129	---	.0010	.0204	$0.0331	.0017	.0007	---	.0698
Refining	.0081	---	.0026	.0265	---	.0013	.0009	.0025	.0419
Disposal of tailings	.0237	---	.0168	.0176	---	.0057	.0045	---	.0683
Assaying	.0245	---	---	.0234	---	---	---	---	.0479
Reagents:									
Cyanide	---	---	---	---	.1543	---	---	---	.1543
Lime	---	---	---	---	.0394	---	---	---	.0394
Lead acetate	---	---	---	---	.0051	---	---	---	.0051
Hydrochloric acid	---	---	---	.0035	---	---	---	---	.0035
General supervision and superintendence	---	$0.0392	---	---	---	---	---	---	.0392
Office	.0296	---	---	.0120	---	---	---	---	.0416
Yarding	---	---	---	---	---	---	---	.0175	.0175
Bullion	---	---	---	---	---	---	---	.0355	.0355
Outside lighting	---	---	---	---	---	---	---	.0016	.0016
By-product	---	---	---	---	---	---	---	.0113	.0113
Flue-dust	---	---	---	---	---	---	---	.0107	.0107
Umpire expense	---	---	---	---	---	---	---	.0020	.0020
Miscellaneous buildings	---	---	---	---	---	---	---	.0139	.0139
Railroad tracks	---	---	---	---	---	---	---	.0071	.0071
Water works line	---	---	---	---	---	---	---	[2] .0118	.0118
Track scales	---	---	---	---	---	---	---	.0003	.0003
Watchmen and fire protection	.0052	---	---	---	---	---	---	.0062	.0114
All insurance	---	---	---	---	---	---	---	.0336	.0336
General expense	---	---	---	---	---	---	---	.0585	.0585
Totals	0.5454	0.0392	0.2551	0.6235	0.2319	0.1090	0.0987	0.3111	2.2139

[1] Power costs $0.0096 per kilowatt-hour.
[2] Water costs $0.01096 per 1,000 gallons.

Table 5. - Distribution of labor for 1929

	Tons of ore treated per 8-hour man-shift			Percent of total labor
	Operation	Maintenance	Total	
Unloading	60.904	(1)	60.904	8.463
Sampling	81.008	359.584	66.111	7.796
Symons crushers	342.288	1,859.456	289.076	1.783
Comminuters	323.005	704.440	221.456	2.328
Screens and elevators .	393.144	860.000	269.804	1.911
Roasters	26.126	247.776	23.632	21.816
Chilean mills	242.256	241.400	120.912	4.262
Conveying	72.067	130.136	46.382	11.113
Amalgamation	240.279	(1)	240.279	2.145
Classification	745.128	(1)	745.128	.692
Leaching	220.749	745.416	170.296	3.025
Slimes	257.357	657.328	184.944	2.787
Filters	307.144	1,349.016	250.184	2.064
Clarifier	845.904	(1)	845.904	.609
Precipitation	328.140	(1)	328.140	1.571
Refining	461.744	(1)	461.744	1.116
Tailings	172.720	742.448	140.120	3.679
Assaying	277.416	(1)	277.416	1.858
General supervision and superintendence	273.376	(1)	273.376	1.886
Power plant	135.522	430.000	103.048	5.001
General expense	37.283	(1)	37.283	14.095

1/ Maintenance included in operation.

Table 6. - Distribution of power for 1929

	Kilowatt-hours per ton of ore treated	Per cent of total power
Sampling, including jaw crushers and rolls	1.2575	4.56
Symons crusher and comminuters	5.1328	18.59
Roasting	4.9574	17.96
Chilean mills	6.7256	24.37
Classifiers, screens, elevators, and conveyors	3.9562	14.33
Amalgamation	.1302	.47
Cyanidation:		
Sand leaching	.3856	1.40
Slime leaching:		
Agitating and settling	1.6241	5.88
Filtering	.5414	1.96
Aeration	.6416	2.32
Precipitation	.0987	.36
Refining	.1442	.52
Disposal of tailings	1.7490	6.34
By-product	.2594	.94
Totals	27.6037	100.00

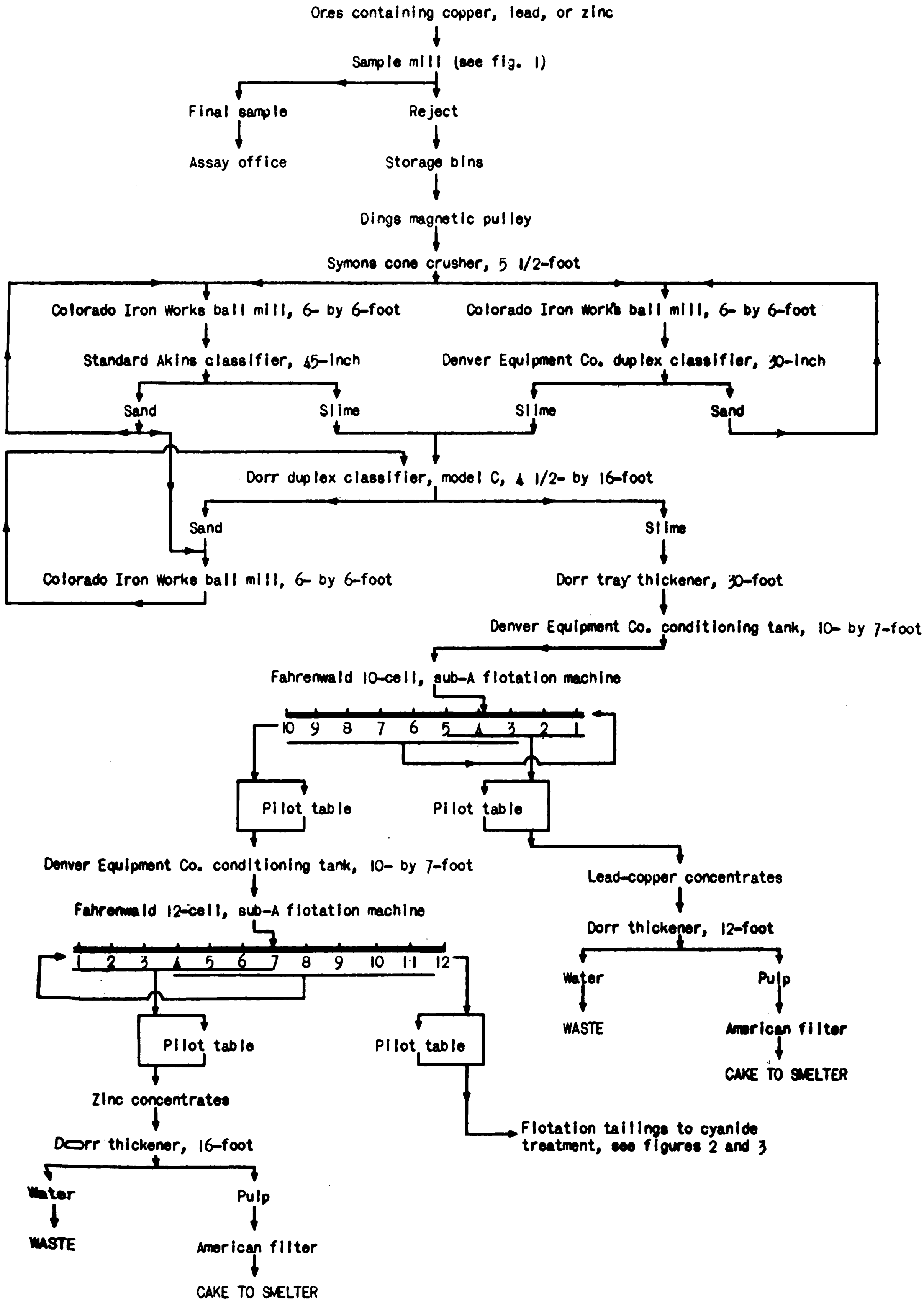
Ores containing copper, lead, or zinc
Sample mill (see fig. 1)
Final sample
Reject
Assay office
Storage bins
Dings magnetic pulley
Symons cone crusher, 5 1/2-foot
Colorado Iron Works ball mill, 6- by 6-foot
Colorado Iron Works ball mill, 6- by 6-foot
Standard Akins classifier, 45-inch
Denver Equipment Co. duplex classifier, 30-inch
Sand
Slime
Slime
Sand
Dorr duplex classifier, model C, 4 1/2- by 16-foot
Sand
Slime
Colorado Iron Works ball mill, 6- by 6-foot
Dorr tray thickener, 30-foot
Denver Equipment Co. conditioning tank, 10- by 7-foot
Fahrenwald 10-cell, sub-A flotation machine
10 9 8 7 6 5 4 3 2 1
Pilot table
Pilot table
Denver Equipment Co. conditioning tank, 10- by 7-foot
Lead-copper concentrates
Fahrenwald 12-cell, sub-A flotation machine
Dorr thickener, 12-foot
1 2 3 4 5 6 7 8 9 10 11 12
Water
Pulp
WASTE
American filter
CAKE TO SMELTER
Pilot table
Pilot table
Zinc concentrates
Flotation tailings to cyanide treatment, see figures 2 and 3
Dorr thickener, 16-foot
Water
Pulp
WASTE
American filter
CAKE TO SMELTER

Figure 4.- Flow sheet of treatment for ores containing lead, copper, or zinc.